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Yu. P. GALCHENKO[1], *Doctor of Engineering Sciences, Leading Researcher*
V. A. EREMENKO[2], *Professor, Doctor of Engineering Sciences, Director of the Research Center for Applied Geomechanics and Converging Geotechnologies, eremenko@ngs.ru*
A. V. MYASKOV[2], *Professor, Doctor of Economic Sciences, Director*
M. A. KOSYREVA[3], *Student*

[1] *Institute of Comprehensive Exploitation of Mineral Resource, Moscow, Russia*
[2] *NUST MISIS, Moscow, Russia*
[3] *Peoples' Friendship University of Russia, Moscow, Russia*

SOLUTION OF GEOECOLOGICAL PROBLEMS IN UNDERGROUND MINING OF DEEP IRON ORE DEPOSITS

The authors present the actual scenario of transition from mining with caving to mining with backfilling in a deep-level ore mine exposed to high horizontal stress of 5H. Backfill mixture is prepared on the backfill preparation plant of the mine using the local material.

Key words: *underground mining, iron ore, backfill, maximum principal horizontal stress, Inelastic strain zones, block, level*

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Introduction

The formation and development of modern civilization was accompanied by outrunning growth of consumption of a wide variety of natural resources.

A special place of mineral and raw materials is associated not only with their determining role in the energy and material support of the development of modern technocratic civilization, but also with the enormous environmental consequences of mining in the form of large-scale and, most often, irreversible disruption of all geospheres of the Earth.

The result of the functioning of natural and technical systems during development of mineral resources changes not only the state of the lithosphere, but there is an extensive system of different influences on biota and ecosystems about natural-territorial complexes in our case — the elements of the natural subsystem.

The technical component of these systems includes the whole complex of actions for extraction from the lithosphere, the part of its substance, which is necessary for the existence and development of modern technocratic civilization. This means that today all of the environmental issues associated with the development of the mineral mining is a direct or indirect consequence of purposeful destruction of certain parts of the lithosphere.

The extensive changes in the original properties of the lithosphere, associated with extraction and lift of minerals to the surface have obvious economic value.

The result is a new object — mining-altered subsoil, which can be considered as an environmental object, and its features used in a Geotechnology are the environmental factors acting on the object.

In this formulation, the problem to determine the basic parameters of a new environmental object, you must give the idea of rock masses in three forms:

- in the undisturbed lithosphere;
- in the zone of technological impact;
- in the area of distortion of geophysical fields (the transition zone properties).

As for the problems of integrated mineral resources management, the state of the stable lithosphere can be interpreted as geological and mining-technical conditions of development of the field. The nature of development processes in the zone of direct anthropogenic impact is fully determined by the type of applied Geotechnology. In the framework of the modern environment, adapting ideas about changes at contacts of biological systems to induced alterations in the subsoil, the zone of distortion of geophysical fields can be interpreted as geophysical ecotone, which is a system having a discrete inner boundary, continuous external boundary and a high gradient of change of properties initiated by the processes of mineral extraction. At the same time, ecological safety of the applied Geotechnology in the lithosphere is provided in the case of minimizing both geophysical ecotone and gradient change of the properties of the array within it [1].

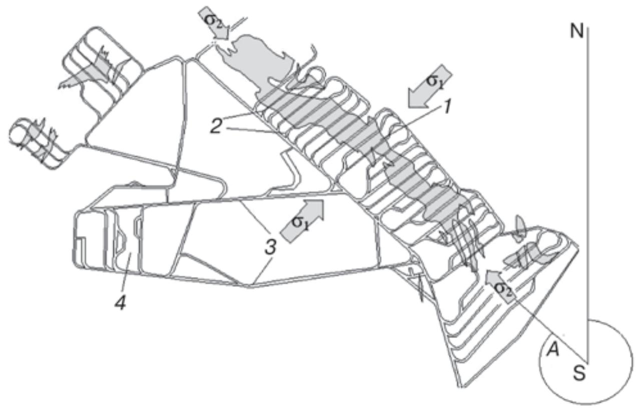

Fig. 1. Tashtagol Mine Level 750 m, plan view: ***1*** **— orebody;** ***2*** **— cross cuts;** ***3*** **— cross measures;** ***4*** **— shaft station; $\sigma_1 = 5\gamma H$ — maximum principal horizontal stress in the stoping area; $\sigma_2 = 3.5\gamma H$ — principal medium normal stress; *A* = 315° — azimuth of the maximum natural compression stress (out of the stoping area)**

It is clear that in this formulation, the only radical solution of ecological problems of the lithosphere in the course of mining is the right choice of applied Geotechnology.

This problem was solved when carrying out research to determine new ways of development of Geotechnology for underground mining at the Tashtagol iron ore deposit.

The Tashtagol iron ore mine is situated in West Siberia, in the area with active modern tectonics. Horizontal stress is 2.5 times higher than vertical stress in the rock mass out of stoping operations and is fivefold the vertical stress in the area

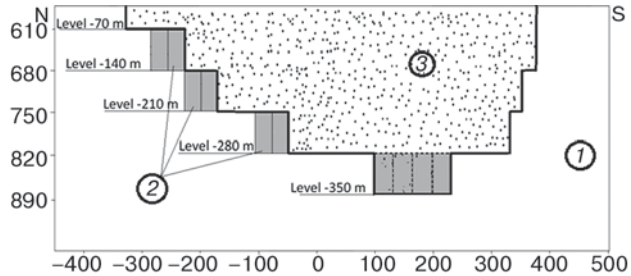

Fig. 2. Schematically, Tashtagol Mine, actual stoping area (side view). Stoping is advanced from the center sideward, concurrently on 4 levels; apart from the bottom level, three levels on the southern side have been mined-out; on the northern side, all the four levels are under mining:
***1* — uncovering blocks (first, block 17 is blasted and extracted, then goes blasting and extraction of blocks 16, 18, 19 etc); *2* — boundary of the mined-out void;**
***3* — mined-out void filled with caved rocks;**
***4* — extraction blocks 1–1, 3, ... 19**

of productive excavation as illustrated in **Fig. 1** [2, 3]. Stoping has reached the depth of 1 km [4]. The deep-level mining undergoes sensibly increased ground pressure [5]. In mining with caving, stoping voids are filled with broken enclosing rock (**Fig. 2**).

The sub-level caving method used in Tashtagol Mine is illustrated in **Fig. 3**. First, a rock block 27–30 m wide and 60–90 m high is prepared (70 m in the case discussed); the width of the rock block equals the to thickness of the orebody that varies from 50 m to 100 m. In the course of the rock block preparation, a haulage cross-cut is driven as well as an undercutting (cushioning) cross-cut over drawpoints and a drilling cross-cut (for vertical blasthole drilling) below the upper haulage level are driven. The block is caved by delayed-action explosion of fans of vertical parallel explosive charges 105 mm in diameter: first the charges in the middle of the block are initiated (delay interval is 0.1–0.1 ms), then marginal charges are set off (delay interval is 0/6–1/0 ms). Blasting is oriented toward the pre-driven cushion cross-cuts (or cushion chambers, their amount is defined by the size of the rock block to be blasted — the thicker is the orebody, the larger is the rock block), toward the balancing room provided by the undercutting level and toward the previously blasted but not excavated neighbor rock block (to prevent from flying-off of rock fragments). Explosive is 100 t to 500 t in weight, which is also governed by the size of the rock block to be blasted; powder factor is 0.3–0.6 kg-t per a block. Ore is loaded from the block bottom to cars by vibration facilities. After ore drawing the mined-out void is filled with caved rocks from above. With this method, the ore loss is 10–12% and ore dilution is 25–30%. Primary concentration method of this black iron ore is magnetic separation. The rock mass is drastically affected by blasting.

At present in the Tashtagol mine, the best part of iron ore occurs in protective pillars. The rest of the reserves are under caving with levels 70 m high. The stoping has approached the boundary of the protective pillars, which complicates geomechanical situation in the mine. Transition to mining with backfilling appears to be infeasible as mined-out voids meant for filling with cemented backfill incur danger of collapse inasmuch as the boundary of caving and the boundary of the protective pillars nearly coincide.

The process flowsheet developed for the transition to mining with cemented backfilling involves formation of an artificial barrier pillar and a separatory ore pillar. The ore reserves in the protective pillars are to be extracted in stopes 70 m high, 27 m long, 13.5 m wide, with mined-out voids filled with cemented backfill.

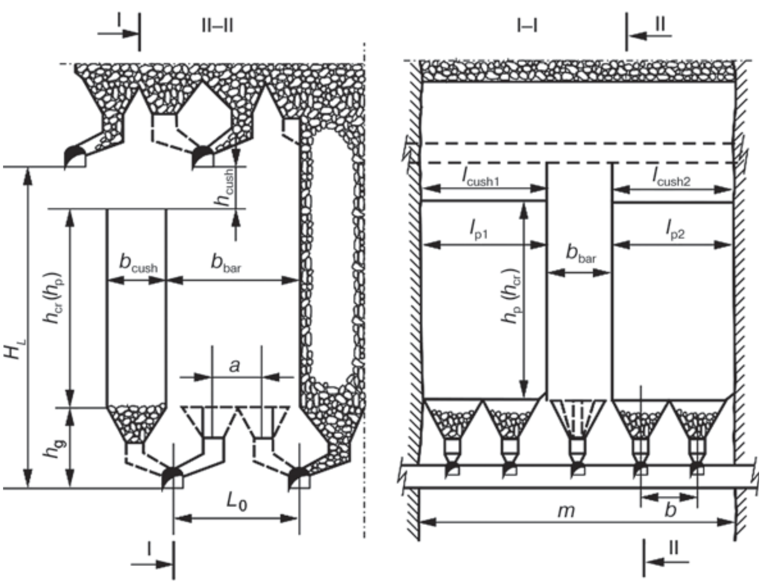

Fig. 3. Layout of sublevel caving:
H_L — height of a level; *m* — orebody thickness; h_b — height of the bottom; h_{cr} — height of the crown; h_{cush} — height of the cushion h_p — height of the pillar; b_{cush} — width of the cushion chamber; b_{bar} — spacing of the blocks (barrier); l_p — length of the pillar; l_{cush} — length of the cushion chamber; L_0 — spacing of the cross-cuts; *a* — spacing of the drawpoints; *b* — spacing of the chambers where ore is loaded to cars by vibration facilities

Transition from mining with caving to mining with cemented backfilling

Locations for the separatory ore pillar and the artificial barrier pillar are designed based on the requirements of stability of the ore pillar and the backfill-made barrier pillar for the stated period of mining, minimum ore loss in the pillars during the transition period, minimum cost of mining with cemented backfilling.

The mathematical modeling using the boundary integral equations allowed understanding of stresses in rock mass on the north side of the orebody and inelastic strain zones that appear in weak and strong rocks as stoping is advanced (**Fig. 4**). The location of the inelastic strain zones was accounted for in development of process flowsheets.

The separatory ore pillar covers ore block no. 2 (see Fig. 5a; block no. 2 is denoted as cross-cut 2, is situated between Level –140 m and Level –70 m. Henceforth, block no. 1 is cross-cut 1 etc, a block includes a few stopes) and ore block no. 1 on Level (–140)–(–70) m

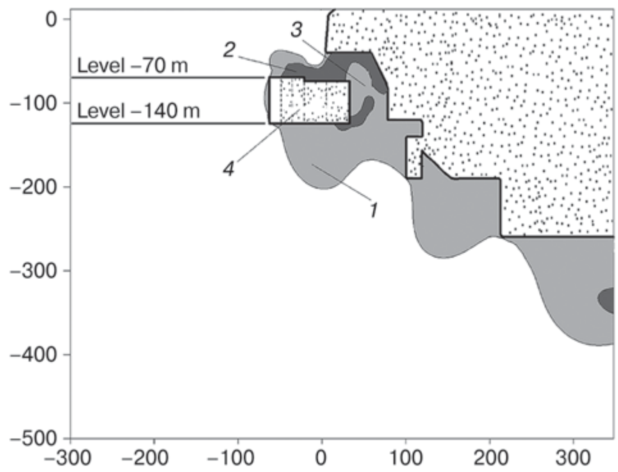

Fig. 4. Inelastic strain zones in influence area of stoping on Level (–140)–(–70) m:
***1* — weak rocks; *2* — hard rocks; *3* — separatory pillar; *4* — stopes; –300–300 m — *X* coordinate; 0–500 m — *Y* coordinate (depth below sea level**

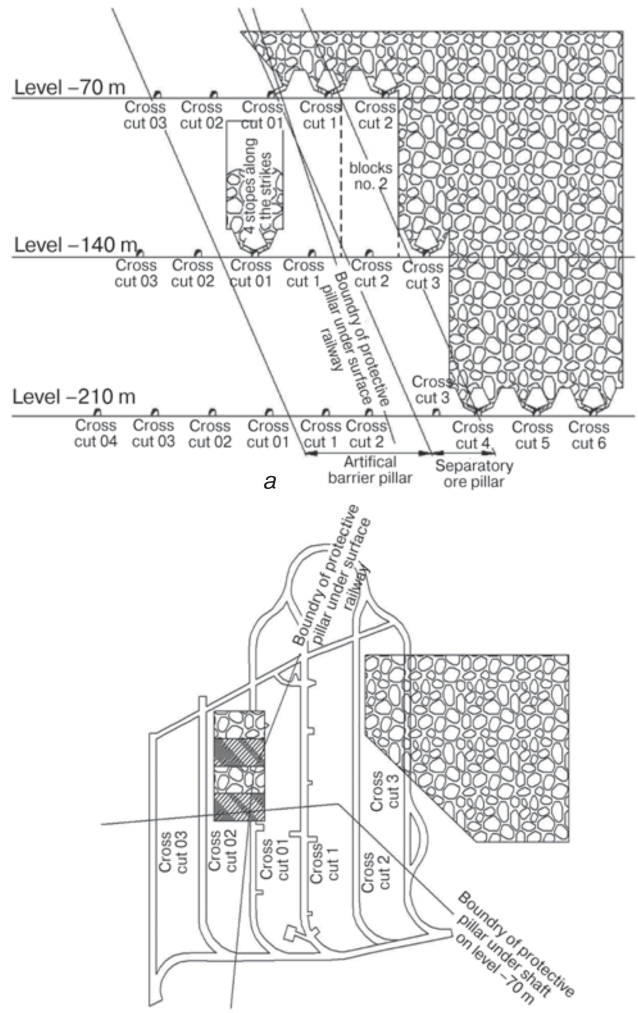

Fig. 5. Transition period stage I:
***a* — profile; *b* — plane view of Level –70 m**

and ore blocks no. 2 and no. 3 on Level (–210)–(–150) m as shown in Fig. 5. The ore in the separatory pillar on Level (–140)–(–70) m is extracted after formation of the artificial barrier blocks 01 and 02 on Level (–140)–(–70) m and artificial barrier blocks 01 and 1 on Level (–210)–(–14) m. Block no. 2 is extracted by caving; block no. 1 is mined with stopes and backfilling. Block no. 1 is mined first.

The artificial barrier pillar 54 m wide is formed in the course of mining in levels, with stopes 70 m high, 27 m long and 13.5 m wide. In block 01 to be mined first, the stopes are arranged so that their long sides are along the strike of the ore body. The artificial barrier blocks 01 and 02 on Level (–140)–(–70) m and artificial barrier blocks 01 and 1 on Level (–210)–(–14) m. Block no. 2 is extracted by caving; block no. 1 is mined with stopes and backfilling. Block no. 1 is mined first. On the south the block 01 perimeter is made concave is order to ensure sustained stability under high ground pressure. The concavity in the center is calculated from the formula [4]):

$$B_C = 0.03m\sqrt{\frac{\sigma}{\left[\sigma_{comp}\right]}} = 0.03 \cdot 67.5\sqrt{\frac{435}{10}} = 7.12 \text{ m}, \qquad (1)$$

where m — ore thickness, m; σ — stress across the strike of the ore body in the stoping area, MPa; σ_{comp} — ultimate strength of the fill mass in three months, MPa.

Block 02 is extracted with stopes arranged so that their long sides lie across the strike of the ore body. In the artificial barrier pillar on Level (–210)–(–140) m, the technology of stoping in block no. 1 is the same is an block 01 on Level (–140)–(–70) m, the stoping technology in block 01 is the same as in block 02, and in block no. 2 — the same as in block no. 1 Depending on the stoping advance rate and efficiency of the stoping with cemented backfill, extraction of block no. 1 on Level (–210)–(–140) m can outtride extraction of block no. 1 on Level (–140)–(–70) m.

The validated sequence of mining on Level (–140)–(–70) m and (–210)–(–140) m is as follows:

Stage I: stoping with cemented backfill in block 01 (**Fig. 5**).

Stage II: extraction of blocks 02 and 03 on Level (–140)–(–70) m (**Fig. 6**).

Stage III: reversed extraction, towards zone of caving on Level (–210)–(–140) m. For the complete extraction of the pillars and making tops of the stopes arrow-headed, location of cross cuts on Level –210 m is shifted by 6.5 m relative to Level –140 m, and the artificial barrier pillar is formed in block no. 1 on Level (–140–(–70) m. The end faces of the adjacent stopes are made so that the surface of the barrier is concaved on the side of the separatory pillar. The concavity is designed to be 7.12 m (**Fig. 7**).

Stage IV: extraction on Level (–210)–(–140) m and formation of the artificial barrier pillar in block no. 1, with arrangement of stopes along the strike of the ore body. After the stoping in block no. 1 has been completed, the artificial barrier pillar is formed in block no. 2 (**Fig. 8**).

The design solutions on mining sequence on Level (–140)–(–70) m and Level (–210)–(–140) m enable: stabilization of geodynamic situation in rock mass before high-rate stoping with cemented backfilling, provision of the design dimensions of the artificial barrier pillar and preservation of the protective pillar.

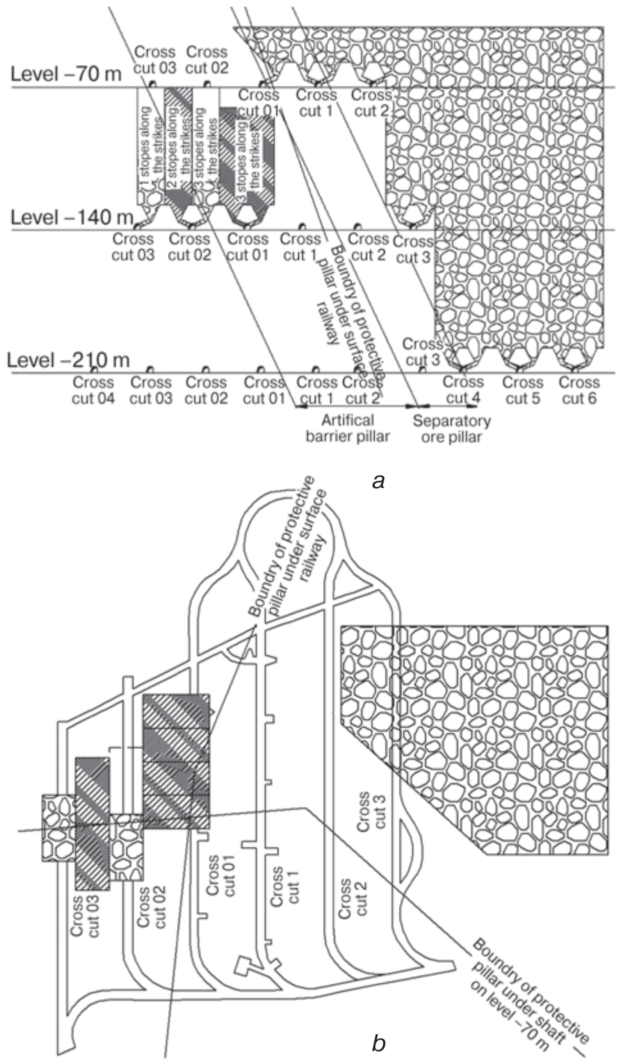

Fig. 6. Transition period stage II:
***a* — profile; *b* — plane view of Level –70 m**

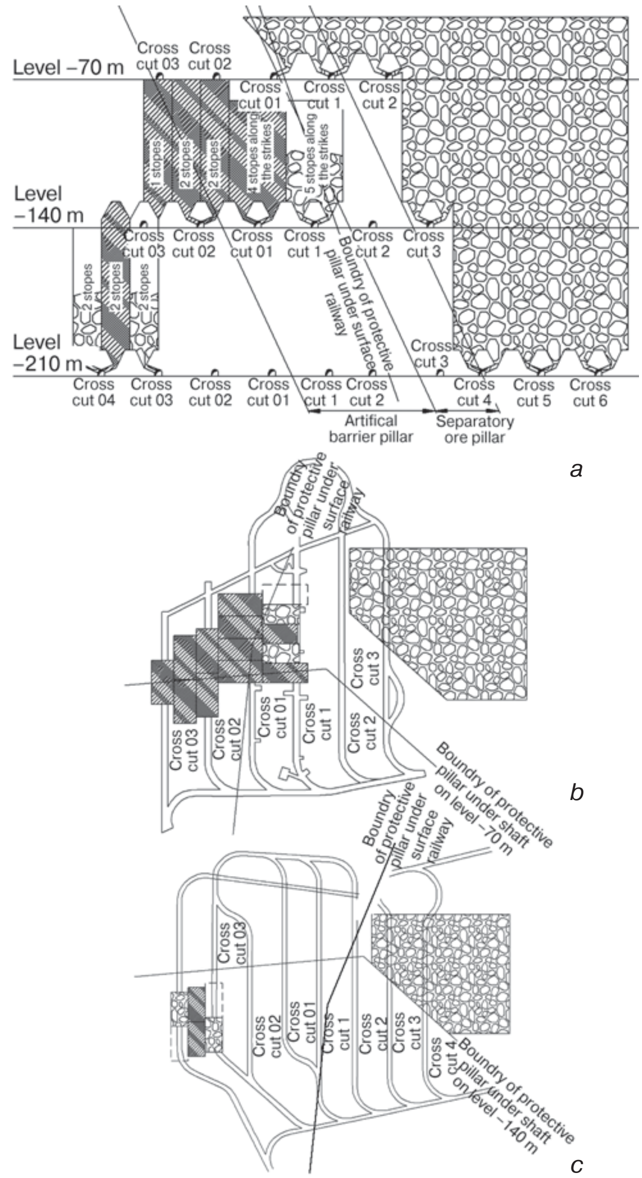

Fig. 7. Transition period stage III:
***a* – profile; *b* — plane view of Level –70 m; *c* — plane view of Level –140 m**

Preparation of cemented backfill in the Tashtagol mine

Profitability of the cemented backfilling technology is governed by the expenditures connected with acquisition of source materials. The cemented backfill is a man-made uniform mixture of a mineral binder, an aggregate material and water. The binder and water, after hydration transformations, become a solidifying mixture that finally passes into petrous state. Introduction of an aggregate (gravel, break stone, sand etc.) into the binder and water mix decreases consumption of the binder, reduces deformation on curing and increases consolidation, density and strength of the backfill owing to mitigating effect of heat-and-moisture-induced conversions during curing.

The Tashtagol mine saves money for purchasing binders. The mine uses granulated furnace slag from metallurgical integrated iron-and-steel works located nearby the mine. The fine fill material is from the closely-spaced quarries (high cost of sand affects the cost of ore mining); the coarse fill material is taken from accumulated tailings of the owned crushing and concentration plant.

To activate hydraulicity of granulated slag and fly ash, the mine uses alkaline activators (Portland cement, lime) and sulfate activators (gypsum and various combinations of anhydrite $CaSO_4$). For better pumping ability and placeability, as well for reduction of spreadability, the backfill is added with plastifiers.

Effect of plastifiers and water amount on flowability of cemented backfill with coarse filler (0–20 mm) is illustrated in Figure 9. Addition of sulphide–spirit distiller's soluble (SSDS) in amount of 0.3% of the weight of a binder greatly improves flowability of the backfill (**Fig. 9**, ***a***). Besides, it is possible to reduce water consumption by 16–20%, or binder consumption, at the preserved design characteristics of the backfill. The SSDS enables higher strength of the backfill already after three months of curing the effect of SSDS is even more pronounced after longer curing. The optimized portion of SSDS addition in a backfill mixture is 0.2–0.25% of the weight of a binder.

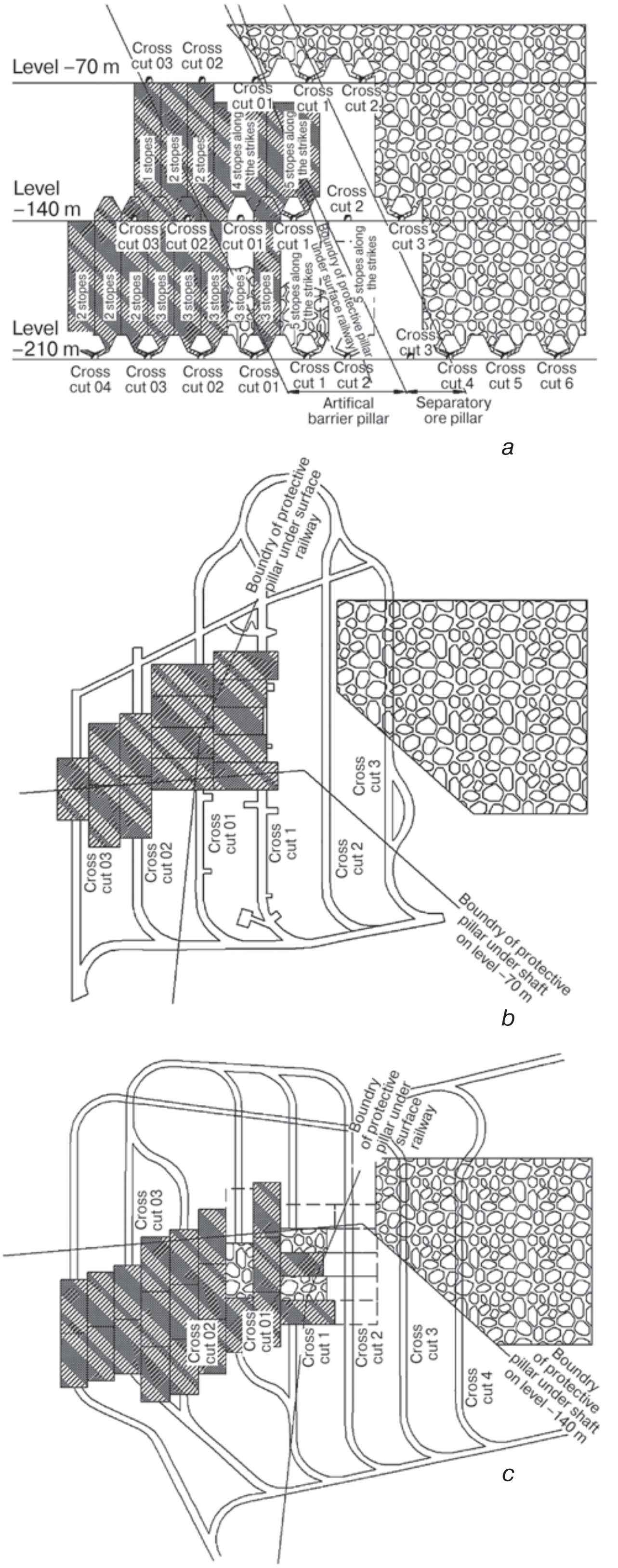

Fig. 8. Transition period stage IV:
a **— profile;** ***b*** **— plane view of Level –70 m;** ***c*** **— plane view of Level –140 m**

Clayey material improves pumping quality of a backfill, prevents segregation, decreases resistance to motion in

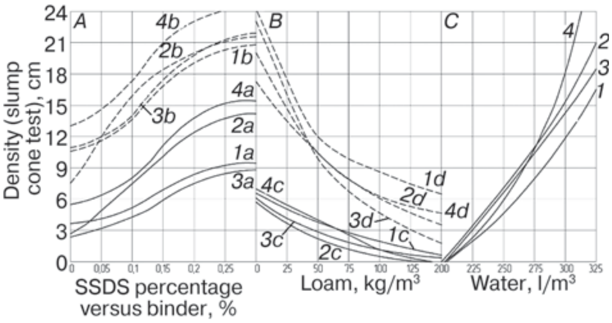

Fig. 9. Flowability of the backfill mixture versus amount of plastifier and water, with the filler 520 mm in size:
1, 2, 3, 4 **— binder 200, 300, 400, 500 kg/m^5, respectively;**
a, b, c, d **— water 240, 250, 270, 300 l/m^3, respectively**

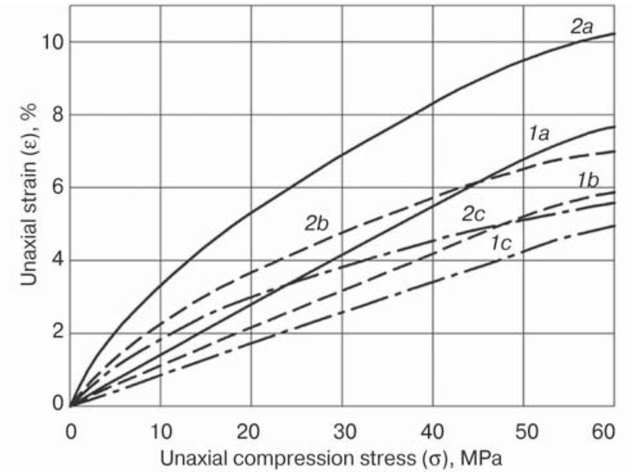

Fig. 10. Straining of the backfill mixture under rapidly growing stress after 28 (*a*), 90 (*b*) and 180 (*c*) days of curing: *1* and *2* are for mixture formulations *1* and *2*

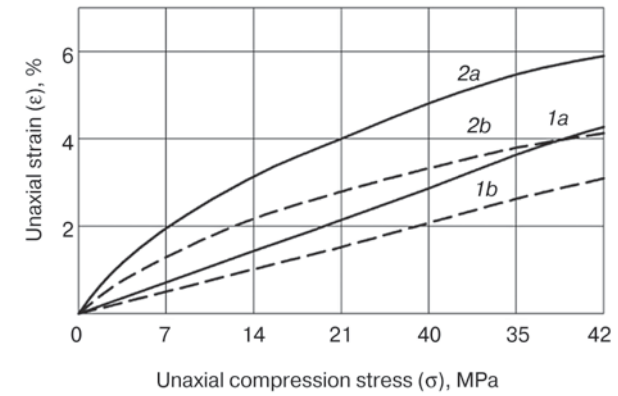

Fig. 11. Straining of the backfill mixture under slowly growing stress after 28 (*a*), 90 (*b*) and 180 (*c*) days of curing: *1* and *2* are for mixture formulations *1* and *2*

pipes and mitigates the pipe wear, but, unfortunately, decreases the backfill flowability (**Fig. 9**, ***b***). The decrease in the backfill flowability takes place with the clayey material addition up to 100 kg per cubic meter. In this case, water consumption grows by 4–5% as compared with the backfill without additions and by 19–20% as compared with the backfill with the addition of SSDS. The issue on adding a clayey material in the backfill mixture or not is decided upon the feasibility study.

The increase in amount of water in the backfill mixture directly influences the backfill flowability, irrespective of the binder consumption (**Fig. 9, *c***). With more water consumed, the backfill is best flowable with the binder consumption not less than 500 kg/m^3 (Fig. 9, *c,* curve *4*).

Backfill placed in underground mined-out voids hardens and incurs pressure from enclosing rock mass [6–13]. As mining goes to deeper levels, load on the backfill grows with time and can reach the stress of an intact rock mass. Different from the enclosing rocks in its physico-mechanical properties, fill mass withstands high compression pressure without failure.

With higher load per unit area, the cemented backfill experiences deformation (**Fig. 10** and **11**). Under rapid continuous compression, the uniaxial strain changes in direct proportion to stress in the backfill mixtures without clayey material addition (Fig. 10, curves *1*). At the stress of 60 MPa the strain curves level-of.

Straining of the backfill without clayey material addition (mixture formulation 2 in Fig. 10) does not obey the direct proportionality, but also levels-off under the stress of 60 MPa.

Strains in the backfill are influenced by the pattern of the backfill loading. When the loading is slow, 0.2 MPa per day (Fig. 11), the strain is lower as compared with the rapid continuous loading growth at the rate of 5 MPa/min (Fig. 10).

The curing time of the backfill in the Tashtagol mine is 90 days. Aimed at attainment of the design strength of 4 MPa and above during 90 days of curing, the cemented backfill consumes 60 kg/m^3 cement and 400 kg/m^3 slag, at the backfill density 2 t/m^3.

The manufacturing parameters of the backfill are: density 1.95–2.2 t/m^3; solid content 86–72%; grinding fineness 70% of 0.14 mm size; pack compression 5%; spreading 5º±1º.

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The pattern and rate of the pressure application on the cemented backfill in the course of ore extraction in the Tahatgol steel thick orebody is under study, in full-scale conditions, using various methods of stress–strain state assessment and ground control.

Conclusions

The designed and validated scenario of the transfer from mining with caving to mining with cemented backfilling in the Tashtagol mine enables stimulation of ore extraction at the designed capacity of the mine.

Owing to the proposed engineering solutions on the sequence of block extraction, geodynamic situation in deep rock mass exposed to high horizontal stress 5γH has been stabilized before high-rate mining with cemented backfill started, the artificial pillar has been formed with the designed dimension, and the protective pillars has been kept stable.

The application of backfilling has become feasible, without the risk of collapse of mined-out voids at the boundary with the protective ore pillars. Finally, the problems of the transition from mining with caving to stoping with backfilling have been successively solved in the mine.

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